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MILLING METHODS AND COSTS AT THE SUPERIOR CONCENTRATOR OF THE ENGELS COPPER MINING CO., PLUMAS COUNTY, CALIF.

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AND COSTS AT THE SUPERIOR CONCENTRATOR OF THE ENGELS COPPER MINING CO., PLUMAS COUNTY, CALIF.[1]

By W. I. Nelson[2]

INTRODUCTION

This paper, which describes the milling practice of the Engels Copper Mining Co., is one of a series on milling methods being prepared for and published by the United States Bureau of Mines.

ACKNOWLEDGMENTS

Acknowledgment is made of the assistance rendered by J. Casagrande, mill foreman of the Engels concentrator, in collecting much of the data presented in this paper.

GENERAL DESCRIPTION

The mill of the Engels Copper Mining Co. is located in Lights Canyon, Plumas County, Calif., 22 miles by railroad from Paxton on the Western Pacific Railroad. The mill treats ore from two mines, the Engels and the Superior, in the proportion of 8 : 2, respectively. The Engels ore is mined entirely by the shrinkage-stope system, the ore being broken to pass 11-inch spaces between grizzly rails. In the Superior mine the pitch of the veins varies from 35 to 85° from the horizontal. The steep veins are mined by shrinkage stopes and the flat veins by the room and pillar system. No timber is used to support the walls in either mine. The ore from the Engels mine is delivered to a concrete mill bin in 6-ton capacity cars, which are dumped by a revolving tipple. The Superior ore is dumped into a steel mill bin by 5-ton capacity skips operated in a vertical shaft.

The concentrator can treat 1,500 tons of ore per day, but averages only 1,000 tons per day owing to the limited capacity of the mines.

Water for milling purposes flows to the concentrator by gravity except during exceptionally cold spells in winter and the dry months of fall; at such periods additional water is pumped from Lights Creek, a lift of 350 feet.

Electric power is purchased from the Great Western Power Co. and this supply is supplemented during the spring and winter months by 400 hp. generated locally by water power.

ORE TREATED

The copper minerals in the ore are principally chalcopyrite and bornite in the average proportion of 3 : 2, respectively. During 1929 the heads averaged 1.524 per cent of copper and from concentrates returns were estimated to contain $0.11 in gold and 0.345 ounce of silver per ton. In the Superior mine, the vein gangue is chiefly a fine-grained tourmaline

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2 - General superintendent, Engels Copper Mining Co., and consulting engineer, United States Bureau of Mines.

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which occurs in a hard quartz monzonite. The chalcopyrite and bornite occur in widely varying proportions scattered through the vein matter. Several veins carry small amounts of tetrahedrite, usually high in silver content, and minor amounts of galena and sphalerite. In the Engels mine the copper minerals occur in altered diorite; the chalcopyrite and bornite are much more intimately mixed than is the case at the Superior mine and occur in an interwoven network of stringers and in fine-grained disseminations through the rock. Magnetite generally occurs through the ore zone in sufficient quantities to cause pieces of ore to be lifted when brought into a powerful magnetic field. Pyrite is present in very small amounts, but does not constitute a problem in flotation treatment.

On account of the fineness of the sulphide particles in the disseminated portions of the orebody it is necessary to grind the ore very fine in order to make a satisfactory extraction and fairly clean concentrates.

HISTORY OF CONCENTRATOR OPERATIONS

A small mill of 150 tons capacity per day was erected at the Upper Camp of the Engels mine in 1914. The capacity was increased later to 400 tons per day by the addition of another grinding unit. It is understood that this mill was one of the first, if not the first, in the United States to practice the all-flotation method of treatment. This plant treated practically all of the oxidized ore from the upper workings of the mine, and copper losses in the tailings were naturally high on this account. Chalcocite occurred in both the sooty and clean steel-gray mineral varieties, neither of which appeared to give any trouble in flotation operations.

In this mill the primary grinding was done by two ball mills; one 8-foot by 36-inch Hardinge, and one 5 by 6 foot Allis-Chalmers. The coarser sands contained in the pulps of the primary mills were further reduced by tube mills.

The ground pulp from the Hardinge unit was treated in four Callow cells which produced finished concentrates and tailings. The tailings were given additional flotation treatment in an 18-inch Minerals Separation machine which produced finished concentrates, middling froths, and finished tailings. The middlings were retreated in the cleaner cells of the Allis-Chalmers unit.

The pulp from the Allis-Chalmers unit was divided between one 12-inch and one 18-inch Minerals Separation machines. The 18-inch machine also handled the tailings of the primary Callow cells of the Hardinge unit. These machines produced concentrates, which were cleaned by a set of Callow cells, and final tailings. The cleaner cells produced finished concentrates and middlings which were returned to the tube mills for additional grinding.

Until 1917, when the Indian Valley Railroad was completed to Engels, the concentrates were trammed down the hill, a distance of 1 mile, and then hauled by truck to Keddie. The transportation of concentrates and supplies by truck during the winter months over dirt roads and mountain grades was a source of much trouble.

In 1918 a new concentrator was completed. This plant, located near the Superior mine and at an elevation which is 1,000 feet lower than the first mill, treated ores from both the Engels and Superior mines. The Engels ore was trammed from the mine to a jaw crusher located near the portal of the No. 6 tunnel. After crushing, the ore was transported to the new mill by an aerial tram.

The grinding unit as first operated in the new concentrator comprised two primary ball mills, one 8 by 6 foot Marcy and one 8-foot by 36-inch Hardinge, with two 6 by 12 foot tube mills for secondary grinding. Owing to the toughness of the ores from the Superior mine and from the deeper workings of the Engels mine the capacity of the Marcy mill operating on coarse feed was far from expectations. A set of rolls was, therefore, installed to reduce

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the size of the feed to this mill. The rolls received as feed the oversize of a trommel having 2-inch holes, and after one pass through the rolls the ore was conveyed to the ball-mill feed bins.

As the mine production increased, two 8-foot by 36-inch Hardinge mills were added to increase primary grinding capacity. When the old mill was dismantled, one 7 by 10 foot ball-tube mill and one 8-foot by 36-inch Hardinge mill were moved to the new concentrator and installed as secondary grinding units. These changes were completed in 1920 and since that time only minor changes have been made in the primary and secondary grinding units. The 7 by 10 foot ball-tube mill was reduced to 6 by 10 foot size by lining the shell with 6 by 6 inch timbers and covering these with cast-iron liners. A model D classifier was moved from a primary Hardinge mill to the 6 by 10 foot tube mill in order to increase the capacity of the latter by operating it with a greater circulating load. The Hardinge mill, from which this classifier was moved, is now operated in circuit with a model C classifier. The model C machine handles the load in a satisfactory manner when operated in open circuit with the ball mill but must be watched closely when operating in closed circuit to guard against overloading. A classifier drainage sump was installed to facilitate the starting up of grinding units after power interruptions.

A concrete ore bin was built upon completion of the No. 10 tunnel and the dismantling of the aerial tram in 1924. The jaw crusher and the No. 6 K Gates crusher were also installed at this time. In 1926 the trommels were replaced by Hum-mer screens. These screens had 1¼-inch openings and the rolls were operated in closed circuit with them. This arrangement produced primary ball-mill feed of 1¼-inch maximum size; this size was the smallest that could be employed with the equipment in service. Mill tests, however, indicated that costs could be further decreased by feeding finer material to the primary grinding mills. As this change would mean a considerable expenditure for the installation of another fine-crushing machine, the plan was never completed.

In 1920 one 18-inch Minerals Separation flotation machine was added to supplement the 24-inch machine originally installed. Later, a 20 by 10 foot surge tank was added in order to maintain a more constant feed to the flotation circuit. A second 24-inch Minerals Separation machine was installed in 1922 to relieve the overloaded conditions of the 18-inch and original 24-inch machines. Scavenger pneumatic flotation cells were also added at this time for additional treatment of the tailings of the Minerals Separation machines.

The original equipment for cleaning rougher flotation middlings and concentrates of the primary machines comprised two pneumatic cleaners which handled middlings and one finishing pneumatic cleaner cell which handled the concentrates. The concentrates of the pneumatic cells handling middlings were recleaned in the finishing machine. The tailings from all pneumatic cleaner cells were returned to the heads of the primary roughers. The amount of sulphide minerals returned to the head of the primary roughers in the tailings products of the cleaner cells was considerably decreased in the latter part of 1921 by installing additional cells to operate in series with the pneumatic cleaners.

The tabulation which follows shows the average tonnage of ore milled per day, the average copper content of heads, the average grade of concentrates produced, the average extraction, and the average milling cost for each year of the period 1920 to 1929, inclusive.

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Average tonnage milled per day, copper contents of heads, grade of concentrates, extraction and milling cost, 1920-1929

Year	Ore milled per day, tons	Assay of heads, per cent copper	Grade of concentrates, per cent copper	Extraction, per cent	Cost per ton of ore milled, dollars
1920	658.20	2.210	29.76	80.93	1.2407
1921	840.70	2.224	29.00	84.48	1.0332
1922	994.40	2.218	28.55	87.18	0.9310
1923	1,050.74	2.134	28.99	88.73	.8314
1924	962.76	2.054	27.89	89.40	.8486
1925	1,134.62	1.979	27.75	93.16	.7460
1926	1,164.70	1.785	24.32	88.66	.6778
1927	1,062.63	1.788	27.47	88.14	.6758
1928	985.90	1.710	27.79	91.58	.6780
1929	1,091.20	1.524	27.76	91.34	.6529

PRESENT METHOD OF CONCENTRATING

A plan showing the general arrangement of machines is presented in Figure 1, and a typical section of the concentrator is shown in Figure 2.

A general summary of method of treatment follows.

The ore is reduced to 1¼-inch size for ball-mill feed by primary and secondary jaw and gyratory breakers followed by rolls which operate in closed circuit with Hum-mer screens.

Grinding is done in two stages by ball mills.

The ground product contains 87 per cent of minus 100-mesh material and is treated in three Minerals Separation flotation machines which produce finished concentrates and three middlings products of different grades.

The middlings are retreated in pneumatic-type flotation machines. These cells produce finished concentrates and middlings which are returned to the head of the flotation circuit.

The concentrates, after dewatering, are loaded into gondola cars and shipped to the Garfield plant of the American Smelting and Refining Co.

The tailings pulp is conveyed in flumes a distance of ¼ mile down the canyon, where the coarser solids are removed. The slime pulp is conveyed an additional 4 miles in flumes and impounded.

Coarse Crushing

The flow sheet of crushing and grinding operations is presented in Figure 3.

The concentrator is provided with two coarse-ore storage bins, one for ore from each mine. Each storage bin is served by a separate coarse-crushing unit. The arrangement is flexible in that ore from one bin may be fed to the crushing unit which serves the other bin.

The size of openings in mine grizzlies is governed by the ability of the rock to flow freely through the chutes which follow the grizzlies. In the case of ore drawn directly from stopes into the mill ore train, occasional pieces are so large that they have to be plugged and either blasted or broken by key and feather before they can be fed into the crusher.

The bin for Superior mine ore is a 24 by 30 foot steel tank having a live capacity of about 500 tons. Ore is drawn through a 5 by 8 foot opening in the bottom of the bin by a 30-inch pan conveyor which feeds one of the coarse-crushing units.

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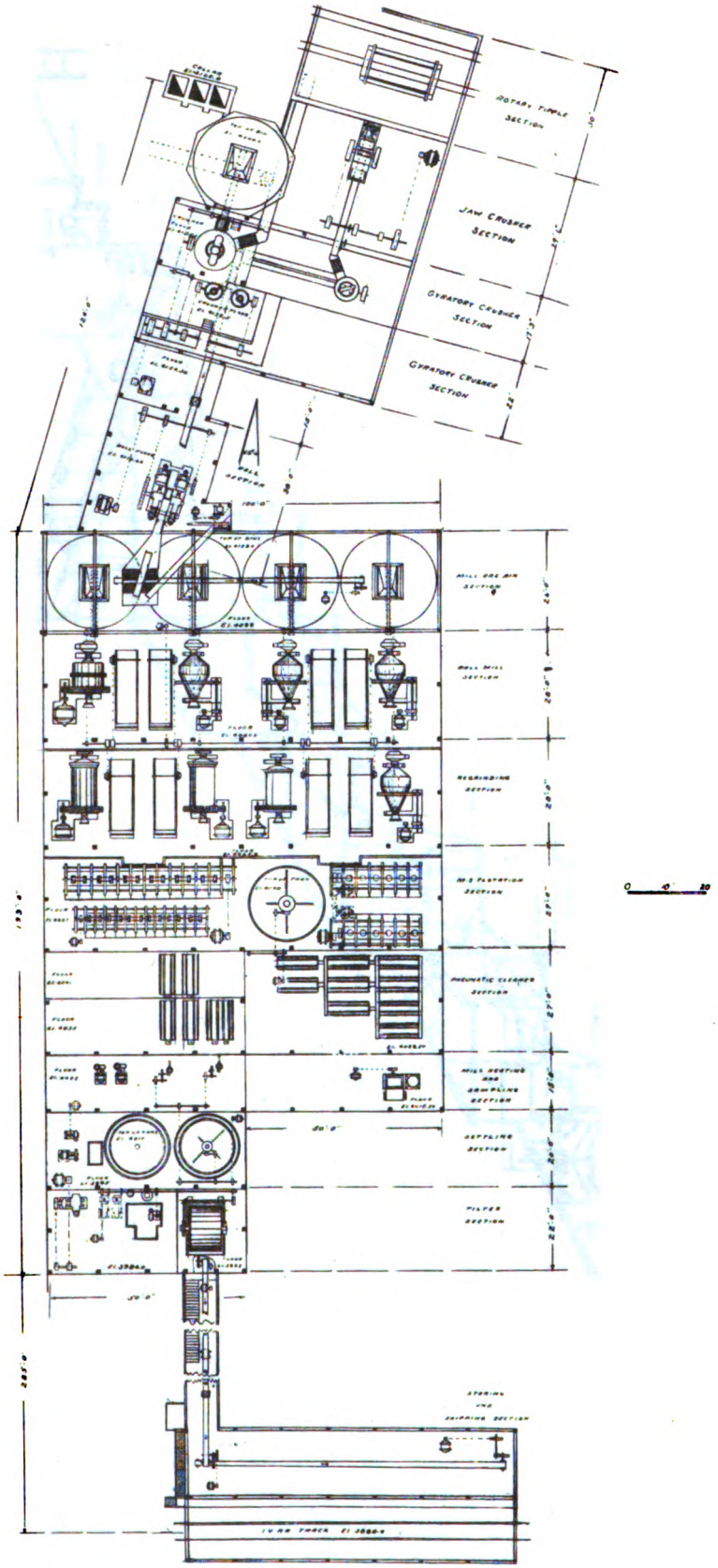

Figure 1.—General arrangement plan of Superior mill

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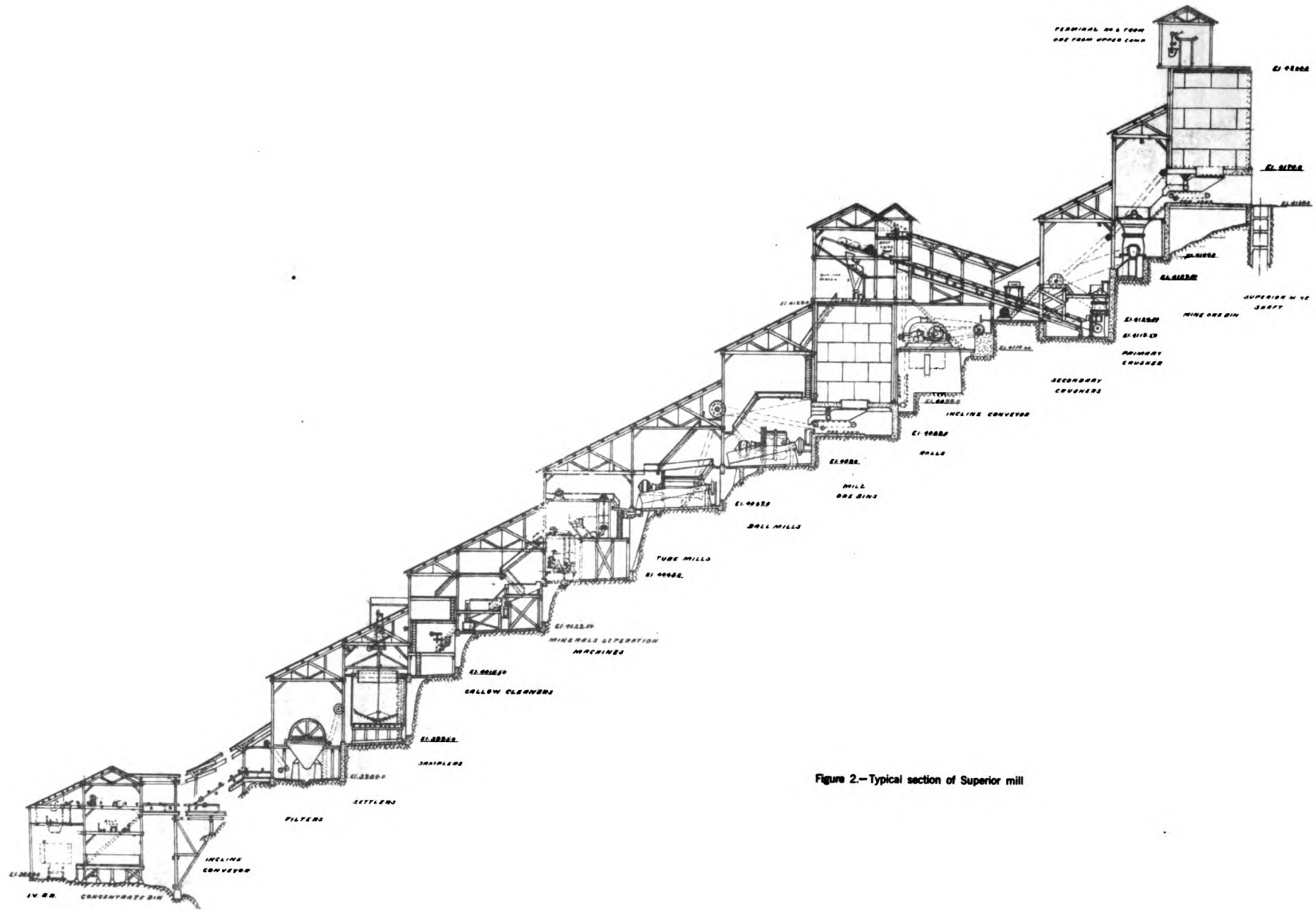

Figure 2.—Typical section of Superior mill

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The storage bin for Engels ore is built of concrete, reinforced by 16-pound rails in the sides and 85-pound rails around the chute openings. The inside of the bin is lined with discarded manganese-steel ball-mill liners to prevent abrasion. The bin has a live capacity for 550 tons of ore. Two side openings are provided which allow ore to be fed to either of the two coarse crushing units. The feed to one of these units is drawn from the bin through an opening, 5 feet wide and 8 feet high, by a rotary feeder. The feed to the other unit is drawn through a corner opening, which is 4 feet wide and 8 feet high, by a pan conveyor. This latter opening is equipped with three 85-pound rails placed across the top and spaced 9 inches apart which makes the net opening 4 feet wide by 5 feet high.

The crushing unit for Superior ore comprises one No. 8 primary Traylor gyratory, which crushes to 3-inch size, followed by two secondary crushers, one a No. 5 Superior McCully and the other a No. 6 K Gates, which crush to approximately 2-inch size. The secondary crushers operate in parallel, each receiving as feed a portion of the primary-crusher product. The three crushers are driven from a line shaft which in turn is driven by a 150-hp. induction motor. When the three crushers are driven from the line shaft without loads the motor requires an input of 25 kilowatts and with full loads 64 kilowatts. The No. 8 primary crusher operates at a speed of 100 r.p.m., the No. 5 McCully at 125 r.p.m. and the No. 6 Gates at 140 r.p.m. The No. 5 McCully and the No. 6 K Gates crushers have capacities of 50 and 65 tons of ore per hour, respectively.

The crushing unit for Engels ore comprises one 24 by 36 inch Blake-type crusher set to crush to 3-inch size followed by one No. 6 K Gates machine which crushes to approximately 2-inch size. The primary-crusher product of this unit can be by-passed to the secondary crushers of the other unit by means of a 24-inch belt conveyor. The Blake crusher is driven at a speed of 222 r.p.m. from a line shaft by two 12-inch, 6-ply belts. At this speed it handles 65 tons of ore per hour. The secondary crusher is driven at a speed of 140 r.p.m. from the same line shaft and can handle 60 tons of primary-crusher product per hour, with the minus 1¼-inch material removed by a grizzly ahead of the machine. The line shaft for this unit is driven by a 150-hp. slip-ring motor. With both crushers operated at full load the motor takes from 90 to 125 kilowatts; without loads it requires 45 kilowatts. With the secondary crusher disconnected and the primary crusher operated at full load the motor takes 65 kilowatts and without load requires 36.5 kilowatts.

Grizzlies, 4 feet wide and with 3-inch spaces, are placed ahead of the primary crushers of both units. The undersize removed amounts to about 20 per cent of the total feed. The manganese-steel grizzly bars are 4 feet long and are set at an inclination of 34° to the horizontal. The secondary crushers are served by grizzlies with 1½-inch spaces.

Both crushing units are operated at the same time when feed for both is available. As previously stated, the No. 8 primary machine can be fed from either ore bin and the ore feed is changed frequently during the shift from one to the other. The average tonnage of ore handled by the entire coarse-crushing plant amounts to 100 tons per hour.

The operating crew for coarse crushing comprises three men per shift. One man operates the feeders and watches the jaw and gyratory crushers in order to keep them running properly. Owing to the wet sticky nature of the ore from the Engels mine, a clean-up man is required to dispose of muck dropped from the return side of the conveyor belts. The third man breaks oversize pieces of ore at the concrete bin openings and also assists the clean-up man. At times the wet sticky ore requires additional labor to keep the plant functioning.

Table 1 gives a screen analysis of the coarse-crushing plant product.

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Intermediate Crushing

The products of both coarse-crushing units join and, after passing over a Merrick weightometer, are fed to two Hum-mer screens which have 1¼-inch holes. These screens operate in closed circuit with a pair of 54 by 24 inch rolls. The rolls receive the oversize of the screens as feed and, after crushing, the product is returned to the screens by a bucket elevator and belt conveyor.

The undersize of the screens provides feed to the primary grinding mills and is distributed into four steel bins, one bin serving each grinding unit. Each bin is 24 by 30 feet, has a total capacity of 850 tons and a live capacity of 500 tons.

The rolls are spaced ¾ inch apart and are driven at a speed of 44 r.p.m. by belts from a line shaft which is in turn driven by a 100-hp. induction motor. Various roll speeds have been tried but when operated at a higher speed than present practice choking resulted which decreased capacity. New roll shells have an outside diameter of 55¾ inches and last from 4 to 6 months. The consumption of roll-shell steel amounts to 0.0716 pound per ton of ore crushed. Longitudinal corrugations form on the shells; these are reduced by grinding with carborundum bricks which are held in position by adjustable arms. When grinding the ridges it is necessary to use a small stream of water applied to the ridge to secure the maximum cutting effect.

One Hum-mer screen handles the product of the coarse-crushing plant and the other is placed in circuit with the rolls. The screens are inclined at an angle of 35° and each has an area of 20 square feet of screen surface. The vibratory magnets are supplied with current by a motor-generator set equipped with a 5-hp. motor. The power input of the motor is 1.5 kilowatts when operated without load and 2.6 kilowatts with load. Screens last about 30 days and magnet coils from 6 to 10 months.

One operator per shift takes care of the rolls, elevator, Hum-mer screens and distribution of the crushed ore to the bins. This operator is also responsible for the proper working of the entire crushing plant in the absence of the shift boss.

Table 1 gives screen analyses and tonnages of the feed, and intermediate and final products of the rolls section.

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Table 1.--Screen analyses and tonnages of feed, intermediate and final products of the rolls section

Screen size	Coarse crushing plant product, new feed to Hum-mer screens, 100 tons per hour, per cent weight	Hum-mer screen oversize from coarse crushing-plant product, 60 tons per hour, per cent weight	Rolls discharge product, 124 tons per hour, per cent weight	Hum-mer screen oversize from rolls product, 64 tons per hour, per cent weight	Undersize products of both Hum-mer screens, 100 tons per hour, per cent weight
Plus 2 inch	24.02	40.03	2.83	5.48	-
Plus 1.50 inch	18.97	31.60	7.94	15.46	-
Plus 1.05 inch	14.84	20.50	30.15	58.53	4.23
Plus 0.742 inch	9.86	7.21	23.10	18.85	22.16
Plus 0.525 inch	7.19	0.50	10.36	0.66	19.33
Plus 0.371 inch	5.45	[1]0.16	7.83	[1]1.12	14.39
Plus 3 mesh	3.46	-	3.54	-	7.85
Plus 4 mesh	2.33	-	3.01	-	6.07
Plus 6 mesh	1.84	-	2.00	-	4.32
Plus 8 mesh	1.59	-	1.59	-	3.56
Plus 10 mesh	0.68	-	1.73	-	2.83
Plus 14 mesh	0.73	-	1.00	-	1.97
Plus 20 mesh	0.85	-	0.63	-	1.63
Plus 28 mesh	0.54	-	0.46	-	1.11
Plus 35 mesh	0.60	-	0.58	-	1.32
Plus 48 mesh	0.69	-	0.49	-	1.30
Plus 65 mesh	0.64	-	0.15	-	0.82
Plus 100 mesh	1.13	-	0.34	-	1.25
Plus 150 mesh	0.31	-	0.25	-	0.42
Plus 200 mesh	1.12	-	0.47	-	1.20
Minus 200 mesh	3.39	-	1.55	-	4.24

1 - Minus 0.525-inch size.

Grinding

There are four primary grinding mills, three of which are 8-foot by 36-inch Hardinge mills and one an 8 by 6 foot Marcy mill. Each mill, as previously mentioned, is served by a separate fine-ore bin. The feed to the grinding mills is drawn from the bottom of these bins and delivered to the mills by conveyor belts. Each primary mill is equipped with a combination scoop feeder and can be operated in either open or closed circuit with the Dorr classifier which receives the ground product. The mills are normally operated in open circuit, but when roll shells are being replaced they are operated in closed circuit.

Each primary mill is followed by a secondary grinding mill which receives as feed the sand product of the primary classifier. Each secondary mill is operated in closed circuit with a model-D Dorr classifier, the classifier sands being returned to the secondary mill. The combined overflow pulps of the primary and secondary classifiers comprise the feed to the flotation circuits.

The primary Hardinge mills operate at slightly different speeds but average 20.5 r.p.m. Each mill is driven by a 150-hp. slip-ring motor, through a 14-inch silent chain, and spur

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gears. The average power input to each motor is 110 kilowatts. The ball load is maintained at about 12 tons by the addition of seven 5-inch balls during each shift. The consumption of balls amounts to 1.032 pounds per ton of new feed. Annealed steel liners are held in place by wedge bars of the same material, and the liner consumption is 0.231 pound per ton of ore milled. A chemical analysis of liner material follows.

	Per cent
Silicon	0.24
Chromium	0.15
Phosphorous	0.028
Manganese	0.62
Combined carbon	0.92
Sulphur	0.051

The tabulation which follows gives the average life of each row of liners and wedge bars over a number of years.

Average life of liners and wedge bars

Location of row	Liners		Wedge bars	
	Row	Life, days	Row	Life, days
Feed end of mill	A	170	-	-
Feed end of mill	B	242	B	158
Cylinder	C	204	C	170
Discharge side of cylinder	D	192	D	178
Discharge side of cylinder	E	408	E	289
Discharge end of mill	F	640	-	-

The mills are stopped and inspected every two weeks of running time to replace thin or worn-out liners. This procedure is strictly followed to prevent undue wear of the mill shell. In spite of this precaution, it has been found that the shell becomes worn by pulp flowing between the ends of the liner rows and also between the "A" row of liners and the shell which necessitates replacement of sections of the shell about every five years of running time. Some experimental work is being done with a rubber lining between the shell and the regular liners, but definite information as to the value of this procedure has not yet been obtained. The combination scoop feeders last about three years and the spur pinions about four years.

The pulp in the Hardinge primary mills is maintained at 80 per cent of solids and each mill has an average capacity of 382 tons of ore per day when operated in open circuit.

The No. 86 primary Marcy mill is driven at a speed of 21 r.p.m. by a 225-hp. slip-ring motor which is coupled to the pinion shaft of the mill by double disks and 10 rubber bushings. The gear and pinion are of the Fawcus type. The power input to the mill is 200 kilowatts. The Marcy mill receives about one-half of the sand product of the primary Dorr classifier in addition to the new ore feed; the balance of the sand from the primary classifier provides feed to the secondary grinding mill of this unit. When operated in this manner the primary mill handles 588 tons of new feed and 200 tons of return sands per day. The mill is operated with a ball load of 12.5 tons and this load is maintained by the addition of twelve 5-inch balls during each shift. The consumption of balls amounts to 1.176 pounds per ton of new feed. Forged-steel and cast-steel balls have been compared over long periods with the

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result that the cast-steel balls showed a lower cost per ton of ore milled. Manganese-steel liners of the block type with corrugated or waved surfaces are used. The liner consumption averages 0.1814 pound per ton of new feed. The pulp in the mill is maintained at about 83 per cent of solids. The feeder scoop lasts about four years and the pinion about two and a half years.

The secondary grinding mills comprise two 6 by 12 foot Power and Machinery Co. ball-tube mills, one Power and Machinery Co. 6 by 10 foot mill which, as previously stated, was converted from a 7 by 10 foot size machine, and one 8-foot by 36-inch Hardinge mill.

Each 6 by 12 foot machine is driven at a speed of 24 r.p.m. by a 150-hp. slip-ring motor through a Falk-type pinion and gear. The pinion shaft is connected to the motor by the overlapping disk and belt-lacing type of coupling. The power input of each motor is 120 kilowatts. The average ball load amounts to 14 tons and is maintained by the addition of 130 pounds of 2½-inch chilled cast balls during each shift. Liners are of chilled iron and are cast with two small lengthwise ridges on each liner. Liner consumption is 0.1052 pound per ton of new feed handled. Pulp consistency in the mills is maintained at 76 per cent of solids. As previously stated, these mills operate in closed circuit with Dorr classifiers. Each mill is equipped with a 2-point feed scoop and carries a heavy circulating load. Pinions last about four years, scoops about five years, and cast-iron screens from six to eight months of operating time.

The 6 by 10 foot mill is driven at a speed of 23 r.p.m. by an old-style 85-hp. slip-ring motor which takes a power input of 96 kilowatts. The ball load is maintained at about 7½ tons by the addition of 80 pounds of balls per shift.

The secondary Hardinge mill is driven at a speed of 17 r.p.m. by a 150-hp. slip-ring motor which requires 75-kilowatts input. The ball load is maintained at about 12 tons by the addition of 100 pounds of balls per shift.

All classifiers used in both primary and secondary grinding circuits are model-DSD Dorr machines, except one model-C machine which is operated with one of the primary Hardinge mills The model-DSD classifiers are 23½ feet long by 6 feet wide and the model-C is 20½ feet long and 4½ feet wide. They operate at a slope of 2½ inches per foot and with rake speeds of 22 strokes per minute. All classifiers and primary grinding-mill feeders are driven from a line shaft which in turn is driven by a 35-hp. motor. The power input to the motor when the line shaft is running and all belts are on idler pulleys is 4 kilowatts. This input increases about 0.8 kilowatts for each classifier operated. It is not believed advisable to reduce the size of the driving motor, since it appears to be well loaded in starting up machines after brief power interruptions. A power interruption of 9 to 10 minutes is usually sufficient to clog a classifier so that it must be drained in order to start. Classifier pulps of the primary Hardinge units are maintained at 50 per cent of solids, 43 per cent in the primary Marcy unit and 45 per cent in the secondary grinding units.

Two men per shift are required for the operation of the primary and secondary grinding circuits. The feed to a primary mill is regulated largely by the sound of the balls in the mill and mills are operated at their maximum capacity as much as possible.

Some interesting facts regarding the operation of a Marcy mill at this plant have been worked out. When the primary Marcy mill was operated in open circuit and all classifier sands were sent to the secondary grinding circuit, the capacity of the Marcy mill was 670 tons per day, but this method of operation required two 6 by 12 foot secondary mills to finish the grinding of the Marcy sands. When the Marcy mill operated in partial closed circuit and handled 588 tons of new feed and in addition 200 tons of return sand, one 6 by 12 foot secondary mill was able to complete the grinding.

The operation of the primary Hardinge mills in partly closed circuit was not tried, principally because of the fact that the ball-tube secondary mills, in series with them, were working at their maximum efficiencies with the primary mills operated in open circuit.

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Table 2 presents typical tonnages and screen analyses of the feed, intermediate, and final products of the primary Marcy mill circuit. Table 3 gives typical tonnages and screen analyses of feed, intermediate, and final products of the secondary-mill grinding circuit which is in series with the primary Marcy unit.

Table 4 presents similar data for the primary Hardinge mill units, and Table 5 for the secondary grinding units which operate in series with the primary Hardinge units.

Table 2.--Typical tonnages and screen analyses of the primary Marcy mill grinding-circuit products

Screen size	Marcy mill new ore feed, 24.5 tons per hour	Classifier sands returned to the Marcy mill, 8.33 tons per hour	Marcy mill discharge, 32.83 tons per hour	Classifier sands sent to secondary grinding as tube mill feed. 12.05 tons per hour	Classifier overflow, 12.45 tons of solids per hour
	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent
Plus 1.05 inch	1.83	-	-	-	-
Plus 0.741 inch	19.45	-	-	-	-
Plus 0.525 inch	18.30	-	-	-	-
Plus 0.371 inch	16.59	-	-	-	-
Plus 3 mesh	9.33	-	-	-	-
Plus 4 mesh	5.99	-	-	-	-
Plus 4 mesh	5.99	-	-	-	-
Plus 6 mesh	4.25	-	-	-	-
Plus 8 mesh	3.49	3.66	2.36	3.71	-
Plus 10 mesh	4.03	5.15	2.97	4.34	-
Plus 14 mesh	1.88	3.00	4.36	6.16	-
Plus 20 mesh	1.53	15.10	7.95	11.01	-
Plus 28 mesh	1.02	20.63	9.00	10.22	-
Plus 35 mesh	1.14	9.09	6.95	13.34	0.02
Plus 48 mesh	1.03	8.58	6.91	12.15	0.73
Plus 65 mesh	0.92	7.33	6.06	9.13	2.09
Plus 100 mesh	1.34	8.49	9.46	10.16	9.48
Plus 150 mesh	0.94	2.29	3.77	2.78	5.72
200 mesh	2.09	4.92	10.85	6.26	19.26
Minus 200 mesh	4.80	7.56	29.36	10.69	62.70

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Table 3.—Typical tonnages and screen analyses of products from the secondary grinding mill of the circuit which operates in series with the Marcy unit

Screen size	Feed to tube mill, 12.05 tons per hour	Tube mill discharge, 50.00 tons per hour	Classifier overflow, 12.05 tons of solids per hour	Classifier return sands to tube mill, 37.95 tons per hour
	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent
Plus 8 mesh	3.71	0.17	-	0.22
Plus 10 mesh	4.34	0.22	-	0.29
Plus 14 mesh	6.16	0.49	-	0.64
Plus 20 mesh	11.01	0.81	-	1.07
Plus 28 mesh	10.22	2.06	-	2.75
Plus 35 mesh	13.34	5.23	-	6.90
Plus 48 mesh	12.15	10.49	0.06	13.81
Plus 65 mesh	9.18	15.00	1.94	19.15
Plus 100 mesh	10.16	18.94	7.33	22.61
Plus 150 mesh	2.78	7.82	5.42	8.58
Plus 200 mesh	6.26	16.05	19.90	14.84
Minus 200 mesh	10.69	22.72	65.35	9.17

Table 4.—Typical tonnages and screen analyses of the primary Hardinge mill grinding-circuit products

Screen size	Hardinge mill new ore feed 15.9 tons per hour	Hardinge mill discharge 15.9 tons per hour	Classifier overflow, 6.85 tons of solids per hour	Classifier sands sent to secondary grinding circuit as tube-mill feed, 9.05 tons per hour
	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent
Plus 1.05 inch	1.66	-	-	-
Plus 0.741 inch	15.91	-	-	-
Plus 0.525 inch	17.90	-	-	-
Plus 0.371	15.25	-	-	-
Plus 3 mesh	11.94	-	-	-
Plus 4 mesh	5.83	-	-	-
Plus 6 mesh	4.09	-	-	-
Plus 8 mesh	3.34	1.44	-	2.63
Plus 10 mesh	3.32	1.94	-	3.41
Plus 14 mesh	2.06	2.44	-	4.28
Plus 20 mesh	1.71	4.35	-	7.65
Plus 28 mesh	1.20	9.41	-	16.39
Plus 35 mesh	2.98	8.52	-	15.00
Plus 48 mesh	2.65	7.36	0.90	12.25
Plus 65 mesh	1.25	7.65	4.79	9.82
Plus 100 mesh	1.61	9.16	12.19	6.88
Plus 150 mesh	0.66	4.90	7.36	3.05
Plus 200 mesh	2.65	11.74	19.97	5.61
Minus 200 mesh	3.99	31.09	54.79	13.03

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Table 5.--Typical tonnages and screen analyses of products from the secondary grinding circuits which operate in series with the Hardinge primary mills

Screen size	Feed to tube mill, 9.05 tons per hour	Tube mill discharge, 40.00 tons per hour	Classifier overflow, 9.05 tons of solids per hour	Classifier return sands to tube mill, 30.95 tons per hour
	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent
Plus 8 mesh	2.63	-	-	-
Plus 10 mesh	3.41	-	-	-
Plus 14 mesh	4.28	-	-	-
Plus 20 mesh	7.65	-	-	-
Plus 28 mesh	16.39	0.56	-	0.71
Plus 35 mesh	15.00	2.43	-	3.14
Plus 48 mesh	12.25	6.16	0.16	7.91
Plus 65 mesh	9.82	10.26	1.81	12.71
Plus 100 mesh	6.88	19.47	7.08	23.04
Plus 150 mesh	3.05	7.52	5.77	8.03
Plus 200 mesh	5.61	22.60	19.82	23.33
Minus 200 mesh	13.03	31.00	65.32	21.08

Flotation

The flow sheet of flotation and dewatering operations is presented in Figure 4.

The overflow pulps from the classifiers of the grinding units flow to a surge tank, which feeds three 12-cell Minerals Separation machines. These machines operate in parallel and produce concentrates froths which are divided into four grades for subsequent treatment. The first cell of each machine produces finished concentrates which go directly to the concentrates thickener. The froth products of cells 2 to 4, inclusive, are cleaned in a Callow-type pneumatic machine referred to as the "finisher" machine, which produces finished concentrates and middlings that join the froth products of Mineral Separation cells 5 to 7, inclusive, for two stages of cleaning in Callow cells. The primary Callow machine of this cleaning unit produces concentrates which join the froth products of M. S. cells 2 to 4 for additional cleaning in the "finisher" machine, and middlings which join the froth products of M. S. cells 8 to 12, inclusive, for treatment in the secondary Callow cleaner of the unit. The froth product of this secondary cleaner is returned to the head of the primary Callow cleaner and the middlings are returned to the surge tank at the head of the Minerals Separation circuit. The tailings of the Minerals Separation machines are given final flotation treatment in six Callow-type cells operated in series as scavenger units. These cells produce froth products, which are returned to the surge tank at the head of the Minerals Separation machines, and tailings which are waste products.

The surge tank at the head of the flotation circuit is 20 feet in diameter and 10 feet high. A vertical cylinder, 2 feet in diameter, is placed in the center of this tank for receiving feed pulp. The bottom of this cylinder is placed 2 feet above the bottom of the tank. Two 6-inch centrifugal pumps, one of which is used as a spare, circulate the pulp from the bottom of the surge tank to a launder which returns the pulp to the top of the surge tank. This launder is equipped with a splitter which deflects a portion of the circulating

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pulp to the flotation machines. The splitter is operated by a float within the surge tank. When the input of pulp to the surge tank increases, the float rises and opens the splitter, thus sending more pulp to the flotation machines and returning less to the surge tank.

The pulp level within the surge tank has a maximum range of 3½ feet. This range gives the flotation operator ample time to adjust the flotation machines when grinding mills are started or stopped and thus prevents sudden changes in the rate of pulp flow through the flotation machines and, in addition, maintains a more uniformly conditioned pulp. Rakes were formerly operated in the bottom of the surge tank but were found to be unnecessary. The circulating pump of the surge tank elevates the pulp a distance of 20 feet to the launder and is driven at a speed of 860 r.p.m. by a direct-connected 25-hp. motor which takes a power input of 11.5 kilowatts.

The Minerals Separation machines, which comprise two 24-inch machines and one 18-inch machine, are of the spitz type. The 24-inch machines are driven by belts from 60-hp. and 75-hp. motors which take power inputs of 50 and 56 kilowatts, respectively. These machines have impeller speeds of 260 r.p.m., and each will handle 550 tons of ore per day when the pulp contains 26 per cent of solids. The 18-inch machine is chain-driven by a 50-hp. motor which takes a power input of 37 kilowatts. The impellers operate at a speed of 315 r.p.m. This machine is of the deep-cell type and will treat 300 tons of ore per day when the pulp contains 26 per cent of solids.

All pneumatic cells are of the Callow type. Six of these cells are operated as cleaners six as scavengers, and three additional scavenger cells are held in reserve as spare units. The scavenger cells are operated in three stages. The first stage contains three cells, the second stage two cells, and the final stage one cell. Each pneumatic cell is 8 feet long and 2 feet wide. The bottom of the cell slopes from a depth of 16 inches at the head end to 40 inches at the foot. Canvas blankets are made of 3-ply material and are stretched over 2 by 1 foot cast-iron pans. The air inlets are located at the bottom of the cell. The air is furnished by a No. 8 Sturtevant positive blower, which operates at a speed of 170 r.p.m. Each cell consumes 135 cubic feet of free air per minute at 4½ pounds pressure, which is equivalent to 1.90 hp.

Two 4-inch centrifugal pumps, belt-driven by 20-hp. motors, handle the pulps in the cleaner flotation circuit. Either pump can be used to lift the return pulps to the surge tank, a vertical distance of 24 feet. When one pump is being repaired, the finisher cell middlings and secondary cleaner cell froths are sent, with the cleaner cell middlings, to the surge tank. The pumps operate at a speed of 1,150 r.p.m. One set of liners usually lasts as long as two runners, which is about 180 days. With one pump elevating pulp to the surge tank and one elevating pulp to the rougher, the power input to the motors is 8¾ and 6.6 kilowatts, respectively.

A fuel-oil collector and a pine-oil frother were used as flotation reagents until the introduction of xanthate. Since the addition of xanthate into the flotation circuit, better results have been obtained. The addition of two kinds of pine oil, added separately to the pulp, has recently been found to give better results than the use of either alone. One of the pine oils makes a brittle froth and the other a fairly tough froth, so that the froth is easily controlled by varying the amounts of oils fed. The use of two frothing oils also made possible the substitution of lime for trisodium phosphate.

The table which follows gives the average amounts and kinds of flotation reagents used with lime and trisodium phosphate, respectively, during 1929.

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Average amounts and kinds of flotation reagents used with lime and trisodium phosphate during 1929

	Pounds per ton of ore milled	
	Using lime	Using trisodium phosphate
Lime	0.1806	-
Trisodium phosphate	-	0.0600
Xanthate	0.0787	0.0753
Fuel oil	0.1045	0.2402
M. S. No. 14 pine oil	0.1571	0.1788
Risor pine oil	0.0485	0.0604
Total	0.5694	0.6147

Lime is crushed to minus ½-inch size and is fed in the dry condition to the ball mills from a hopper by a rotary feeder. The lime feeder is driven from the conveyor which feeds the grinding mills. The amount of lime fed is enough to produce an alkaline pulp with phenolphthalein indicator. The pH value of the pulp water amounts to about 8.2.

When trisodium phosphate was used a batch of the latter was charged into a mixing tank daily and the resulting solution was introduced at the ball mills. The amount fed was determined by trial, the lower limit being indicated by erratic flotation results.

Xanthate solution, which contains 5 per cent of xanthate by weight, and all oils are stored in 100-gallon capacity tanks, and pipe lines connect these tanks to the various points of feeding.

Fuel oil is fed through a petcock to a small plunger pump which in turn feeds this reagent to the primary ball mills. Xanthate is added to the pulp at the head cell of each Minerals Separation machine, and a very small additional amount is added to the pneumatic finisher cell. The tough frothing pine oil is fed to the head cell of each Minerals Separation machine, and small amounts of the brittle frothing pine oil are added at the same points and to the seventh cells as needed.

The reagent tanks are equipped with floats and gages which give the operator a quick check on the consumption of reagents. The additions of reagents are governed entirely by the appearance of the froth.

Pulp density in the Minerals Separation machines is maintained between 26 and 28 per cent of solids. The dilution of pulp encountered between the classifiers and the M. S. machines is due to the return middlings pulp. Water must be added to break down the froth and also to dilute the middlings in order to obtain a satisfactory grade of concentrates in the cleaner cells. The middlings pulp returned to the surge tank averages 7.5 per cent of solids. Pulp densities are determined twice per shift by the flotation operator. The pulp is placed in a conical container and weighed on a vender's-type spring scale equipped with a graduated scale which gives direct density readings.

The flotation concentrates average 32 to 34 per cent of insoluble. The reduction of insoluble content without appreciable loss of copper can be accomplished only by additional grinding and floating.

The Minerals Separation machines produce 64.94 per cent of the total concentrates, which contains 66.70 per cent of the total copper recovered. The cleaner cells are operated from the viewpoint of removing the maximum amount of sulphides from the circuit, consistent with suitable grade, before returning the pulps to the Minerals Separation machines. The loss of copper in the final tailings varies directly with the copper content of the return middlings.

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From data obtained in this plant it has been found that the pneumatic cells give much better results than mechanical cells when operated as cleaners, owing to greater flexibility and ease of control of the former.

Table 6 gives typical amounts and copper contents of the concentrator feed, the intermediate, and final products. Table 7 presents sizing assay tests of typical concentrator heads, concentrates, and tailings and also gives indicated recoveries of copper for each screen size based on heads and tailings assays.

Table 6.—Typical amounts and copper contents of concentrator products

Product	Weight, tons per 24 hours	Copper, per cent
New ore fed to Minerals Separation machines	1,091.00	1.54
Finished concentrates from No. 1 cells of M. S. machines	33.57	29.18
Concentrates from M. S. cells 2 to 4, inclusive, sent to finisher cell	39.75	13.42
Concentrates from M. S. cells 5 to 7, inclusive, sent to primary cleaner	43.43	3.13
Concentrates from M. S. cells 8 to 12, inclusive, sent to secondary cleaner	34.56	1.92
Tailings from M. S. machines, sent to scavenger cells	1,055.98	0.16
Finisher-cell concentrates, finished concentrates	18.12	27.00
Finisher-cell tailings, sent to primary cleaner	53.16	12.18
Primary-cleaner concentrates, sent to finisher cell	31.53	19.98
Primary-cleaner tailings, sent to secondary cleaner	76.40	3.37
Secondary-cleaner concentrates, sent to primary cleaner	11.34	11.22
Secondary-cleaner tailings, sent to head of M. S. machines	99.62	1.96
Scavenger-cell concentrates, sent to head of M. S. machines	16.67	1.05
Scavenger-cell tailings, final waste product	1,039.31	0.15

Table 7.—Typical screen analyses of concentrator heads, concentrates, and tailings

Screen size, mesh	Heads		Concentrates		Tailings		Indicated recovery based on heads and tailings, per cent
	Weight, per cent	Assay, per cent of copper	Weight, per cent	Assay, per cent of copper	Weight, per cent	Assay, per cent of copper	
Plus 65	2.80	0.47	0.35	7.86	2.57	0.28	45.31
Minus 65 plus 100	10.79	0.53	4.52	10.43	11.03	0.21	59.50
Minus 100 plus 150	4.91	0.82	3.55	11.60	5.10	0.15	81.00
Minus 150 plus 200	19.08	1.20	17.50	18.00	20.54	0.12	89.23
Minus 200	62.42	1.94	74.08	32.90	60.76	0.15	94.73
Composite	100.00	1.55	100.00	28.41	100.00	0.15	90.08

DEWATERING OF CONCENTRATES

The finished concentrates pulps from the flotation machines, which contain 37.5 per cent of solids, flow into a 16 by 16 foot Dorr thickener. This thickener has a froth retaining

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wall built for a distance of 2 feet above the top of the tank. The overflow of the thickener goes to a 16-foot diameter settling tank which is equipped with a conical bottom; the overflow of this tank is returned to the surge tank at the head of the flotation machines. The thickener underflow, which contains 75 per cent of solids, is discharged periodically into the filter. The thickener rakes are driven at a speed of 1 revolution in 2½ minutes by a 7½-hp. motor which requires a power input of 2.35 kilowatts. A 12-foot derrick is built over the thickener drive wheel, and the rakes can be raised as much as 7 feet by a set of 3-ton chain blocks which are suspended from this derrick.

The 11½ by 8 foot Oliver continuous filter is operated at a speed of one revolution in 7½ minutes. The feed pulp is heated by live steam to a temperature of 80° C., as it has been found that without this heating of the pulp, the filter cloth becomes so badly choked that it is useless after a run of one week's duration. This choking takes place even after steaming and scrubbing the cloth twice each day between shifts.

An average vacuum of 21 inches of mercury is maintained by a Doak 14 by 8 inch reciprocating pump which operates at 250 strokes per minute. A 15-hp. motor drives the filter and the vacuum pump from a line shaft and requires a power input of 5.72 kilowatts when operated under load. A "blow" of air is given to each filter section as it reaches the scraper and is maintained for two sections below the scraper. The filter cloth is made of loose twilled threads and dewaters 12,000 tons of concentrates or 416.64 tons of concentrates per square foot before being discarded. The life of the cloth is dependent on the life of the winding wire, as the scraper blade cuts the cloth when the wire wears thin.

The filter is operated on two shifts per day and has capacity to dewater from 3 to 10 tons of solids per hour, according to the amount of slimes contained in the pulp. This capacity amounts to from 0.00108 to 0.00347 ton of solids per square foot of filter area. The moisture content of the filter cake averages between 10 and 11 per cent.

DEWATERING AND DISPOSAL OF TAILINGS

The tailings plup flows in a flume to one of two dewatering plants. These dewatering plants, termed "upper" and "lower," are used alternately, the former being used principally during the summer months.

The flume which conveys the tailings to these units is 4,600 feet long and is set at grade of ¼-inch to the foot. The bottom of the flume is 5½ inches wide and the sides have a slope of 4 inches per foot vertically.

The upper dewatering unit is equipped with two model-C Dorr classifiers which remove the coarse material. The classifier sands are conveyed by launder to a bucket elevator which in turn discharges the material to a V-bottom flume. The flume, which is set with a grade of 3 inches per foot, conveys this pulp to the coarse-tailings reservoir.

The lower dewatering unit is similar to the upper except that three 6 by 20½ foot Akins classifiers are used for separating the coarse material from the pulp.

The classifier overflow pulps of either dewatering unit go to an 80-foot Dorr thickener. This thickener produces a clear overflow which is discharged into Lights Creek and an underflow which is conveyed by flume for a distance of 4 miles to the slime-storage dam.

The Dorr classifiers of the "upper" unit operate with a rake speed of 14 strokes per minute and with a bottom slope of 1½ inches per foot. The consistency of the sand pulp is controlled by adding or removing cross pieces which are fitted to the classifier sand discharge.

The bucket elevator which delivers the sand product to the flume is 38 feet high and is equipped with a 12-inch 7-ply rubber covered belt which is driven at a speed of 345½ feet per minute by a 10-hp. motor. This belt operates over head and tail pulleys which are 30

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inches in diameter and is equipped with 10 by 6 by 6 inch buckets spaced 7 inches apart. Two ½-inch holes are drilled in the bottom of each bucket to prevent the sand from settling and sticking.

The Dorr thickener shaft is driven at a speed of 1 revolution in 13 minutes by a 7½-hp. motor. Lime is added to the thickener feed pulp as it leaves the drag classifiers in sufficient amount to produce a clear overflow at the thickener. The lime feeder consists of a 2-cubic foot capacity hopper equipped with a 4 by 6 inch reciprocating plunger which operates across the bottom. The plunger has a 6-inch opening and discharges 140 cubic inches of freshly slaked lime every 5 minutes. The feeder discharges the slaked lime into a 4 by 4 foot mixing tank. A stream of water flows continuously into this tank, and the charge is mixed by a revolving iron weight supported at the end of a shaft. The amount of lime fed per shift is governed by the frequency with which the feed hopper is filled. The average amount fed is 0.614 barrel per 100 tons of tailings.

The reservoir for coarse tailings has an area of 14 acres. The retaining dam is built of sand by a 30-hp. caterpillar and a 1½-yard self-dumping scraper. This dam is 6 feet wide at the top and is usually maintained from 2 to 5 feet higher than the level of the settling pond. The top of the tailings pile slopes about 2 inches per foot in a direction away from the dewatering plants.

Cross dams are built every 300 feet and these serve as approaches by which the tractor and scraper travel on and off the outside dam. The dam has an earth core about 8 feet high and the sand dam is built to a height of 60 feet above this core. A toe of mine rock, 20 feet high, supplemented by log cribbing protects it from the high water of Lights Creek. The earth core is impervious to water, and this caused trouble during the winter months by making a plastic mass of the sand, which caused portions of the dam to slough off. This difficulty was avoided by driving a number of 2-inch pipes, having screened well points at the ends, into the dam at points low enough to maintain the water level below the earth core. No sloughing has been experienced since the introduction of this method of draining. It has been also found necessary to build up the dam for winter storage before freezing weather sets in. Constant attendance is required only during the spring thaw when the water from melting snow and ice washes the sand rapidly down to the dam and causes choking in the overflow boxes, if not watched. Brush is placed along the top of the dam on the prevailing wind side to prevent wind cutting. The height of water in the pond is maintained as low as possible consistent with obtaining a clear overflow. During the summer months there is very little overflow water owing to evaporation, but the condition is otherwise during the winter months, and lime is required for settling. One man with a tractor can build about 300 cubic yards of dam per shift.

The storage dam for slime controls an area of 47 acres. This area is divided into four sections by cross dams. When a section has filled to such a height that it becomes necessary to raise the dam, the slime pulp flow is changed to another pond. It requires from four to six months after this for the slime to dry sufficiently to allow a soraper to be used over it.

SAMPLING

A sample of the heads is taken by a mechanical sampler from the classifier overflow pulp just before it reaches the surge tank at the head of the flotation machines. The cutter crosses the pulp stream every 14 minutes, and when the concentrator is treating 1,000 tons of ore per day, it cuts a sample which contains about 4 pounds of solids per shift.

The tailings sample is obtained as the pulp enters the main flume. The cutter crosses the pulp stream every 9 minutes and obtains a sample which contains about 3½ pounds of solids per shift.

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A sample of concentrates is obtained just as the pulp enters the thickener. The pulp stream is cut every 7 minutes and the weight of solids in this sample per shift amounts to about 4 pounds. A grab sample of concentrates is also taken periodically from the filter cake by the operator for checking purposes. A third concentrates sample is obtained from the gondola. After a gondola is loaded the top is leveled and borings obtained at 12 points with a 1¼-inch auger. The samples are taken at regular intervals in two rows along the length of the car. This sample is used for determinations of moisture, copper, silver, gold, and insoluble contents.

The cutters of all mechanical samplers are actuated by weights. This method of operation is used to prevent the spoiling of samples by power interruptions which cause the cutter to stick in the stream.

Samples are dried each shift on steam hot plates and sent to the assay office each morning. Assay returns are reported to the mill usually before the day shift is over. A sizing test is made on the head sample of each shift to keep a check on classifier operation. The concentrates sample is assayed for insoluble content in addition to copper to aid the operator in judging the grade being produced.

WATER SUPPLY

As previously stated, sufficient water for concentrator operations is usually available from surface sources. This water flows by gravity to a 50,000-gallon capacity storage tank. When a shortage of surface water develops, the deficiency is made up by using mine water from the Engels mine. When mine water is used, trisodium phosphate must be fed to the ball mills to insure uniform results in flotation, as the mine water, although not acid, apparently contains impurities which affect flotation operations adversely. The consumption of new water amounts to 680 gallons per ton of ore milled.

METALLURGICAL DATA

Table 8 presents metallurgical data for the year 1929.

Table 8.—Metallurgical data for the year 1929

Total ore milled	tons	395,042
Operating time per day	hours	24
Average ore milled per 24 hours	tons	1,091.30
Total concentrates produced	tons	19,813.08
Average concentrates produced per 24 hours	tons	54.73
Recovery of total copper	per cent	91.34
Ratio of concentration	tons into 1	19.94
Consumption of 5-inch balls per ton of ore milled	pounds	1.104
Consumption of 2½ inch balls per ton of ore milled	pounds	0.72
Total ball consumption per ton of ore milled	pounds	1.824
Liner consumption per ton of ore milled	pounds	0.32
Insoluble content of concentrates	per cent	34.52
Material plus 100-mesh size	per cent	12.64
Average lime used per ton of ore to settle tailings	barrels	0.00614

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CONVEYING

The No. 8 gyratory crusher is fed, as previously stated, from either of two ore bins, each of which is served by a pan conveyor. The shock of dumping rock into an empty bin is taken up by swing hammers. One conveyor, which is 30 inches wide, 12.5 feet long, and which operates at a speed of 4.4 feet per minute, draws mine ore from the bottom of the steel bin. The other, which is 36 inches wide, 36.5 feet long, and which operates at a speed of 7.5 feet per minute, draws mine ore from the corner of the concrete bin. The pans of each conveyor have 1½-inch wood fillers which are covered by mild-steel plates 1½ inches thick. Each conveyor operates at a capacity of 90 tons of ore per hour.

The Blake-type crusher is fed from the concrete bin by a 30 by 60-inch rotary feeder which operates at a speed of 0.455 r.p.m. The storage-bin discharge opening is equipped with five 4 by 6 inch fingers which permit barring without gushing of ore when the opening becomes choked. These fingers are lined with manganese-steel plates where contact is made with the ore and are otherwise similar to the fingers used in underground chutes. The crushed product of the Blake machine drops onto a 24-inch rubber-covered conveyor belt which travels at a speed of 302 feet per minute. This belt is 75 feet long and elevates the ore a distance of 5 feet. With a 3/16-inch rubber cover it has a life of about 720,000 tons of ore.

The pan conveyors, rotary feeder, and the conveyor which handles the product of the Blake crusher are driven from line shafts which drive the respective crushers.

The products from the No. 5 McCully and the No. 6 K Gates crushers are conveyed to the Hum-mer screens by a 24-inch rubber-covered belt which travels at a speed of 285 feet per minute. This belt is 180 feet long, passes over a Merrick weightometer, and elevates the ore a distance of 28 feet. It is driven by a 10-hp. motor which requires a power input of 4.4 kilowatts without load and 6.8 kilowatts at full load. With a 3/16-inch rubber cover the life of this belt is 1,400,000 tons of ore.

The oversize from the Hum-mer screens is fed to the rolls by a 28-inch belt which is 6 feet long and which travels at a speed of 14.5 feet per minute. It is equipped with side boards, 2½ feet high, to hold the material transported. It has a life of about 900,000 tons or ore.

The roll product is returned to the Hum-mer screens by a bucket elevator and conveyor. The bucket elevator is 55 feet high and slopes 80° from the horizontal. It is equipped with a rubber-covered 10-ply 28-inch belt and buckets 20 by 8 by 8½ inch spaced 14 inches apart lip to lip. The belt is driven at a speed of 306 feet per minute by a 20-hp. motor which requires a power input of 8.5 kilowatts without load and 17 kilowatts with full load. This elevator will handle 150 tons of ore per hour, and its belt has a life of 1,700,000 tons of ore.

The 22-inch conveyor which receives the roll product from the elevator and delivers it to the Hum-mer screens is 30 feet long and is driven at a speed of 245 feet per minute by a 5-hp. motor which requires a power input of 1.5 kilowatts without load and 4.6 kilowatts with full load. It has a life of about 1,700,000 tons of ore.

The undersize product of the Hum-mer screens is transported to the ball-mill feed bins by a 16-inch rubber-covered belt. This belt is driven at a speed of 310 feet per minute by a 10-hp. motor which consumes 3.5 kilowatts without load and 5.5 kilowatts with full load. The average life of this belt is 800,000 tons of ore. It is equipped with a movable distributor for unloading and delivering the ore to the desired bin.

The ball mills are fed by 24-inch belts equipped with covers ⅛-inch thick. The feeder belts which serve the Hardinge mills operate at a speed of 13 feet per minute and have a life of about 400,000 tons of ore. The Marcy-mill feeder belt travels at a speed of 22 feet per minute and also has a life of about 400,000 tons of ore. Two of the ball mills were originally fed by 18-inch pan conveyors. The upkeep of these pan conveyors was so high that rubber belts were substituted when the pans wore out.

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Concentrates are transported down to the railroad cars by three belts arranged in series The first belt is 14 inches wide by 155 feet long, has a drop of 80 feet, and travels at a speed of 150 feet per minute. Due to the steep slope of the belt, it is necessary to enclose it with boards to keep the concentrates from rolling off. Sacks are suspended every 6 feet from the underside of the top cover to further check this rolling action. The second belt is 12 inches wide, 170 feet long. and operates at a speed of 66 feet per minute. This belt is operated at sufficiently low speed to give it the proper load per square foot for the operation of the weightometer over which it passes. The concentrates pass finally to the third belt which runs at right angles to the second belt and passes over the railroad siding. This belt is 75 feet long and travels at a speed of 175 feet per minute. Concentrates are scraped from it by wedge-shaped splitters and drop directly into the gondolas for shipment to the smelter. The weightometer is equipped with a cam mechanism which rings an electric bell for every 3.3 tons of product carried by the belt. This bell notifies the attendant to move the splitter so as to obtain an even distribution over the car. The concentrate conveyors have a life of about 78,000 tons of concentrates. They are driven by two 5-hp. motors which requir a power input of 2.47 and 1.65 kilowatts, respectiely.

The heavy machinery of the crushing plant is handled by crawls and manually operated chain blocks. Te rolls are served by a short crawl.

Repair parts for the remainder of the mill are handled with long slings and chain blocks suspended from the trusses or skidded into place.

Under the conditions noted, the follwing tabulation gives the number of men required and the time necessary to make repairs as listed.

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Time and men required to make repairs

Nature of repair	Time required, hours	Number of men required
Crushers:		
Renewing jaw plates	32	6
No. 8 Traylor crusher:		
Babbitting eccentric	24	2
Installing eccentric	8 to 12	6
Putting concaves in barrel	24	2
Changing barrel	16	8
Mantle on shaft	8	6
Changing shaft	12 to 16	8
No. 6 K Gates crusher:		
Babbitting eccentric	8 to 12	2
Installing eccentric	6	6
Changing barrel	8	6
Mantle on shaft	8	6
Changing shaft	6	6
Rolls:		
Shrinking shells	12	5
Changing rolls	12 to 36	9
Babbitting roll bearings	24	6
Elevators:		
Punching belt	12	2
Installing buckets	12	7
Hum-mer screens:		
Changing one screen	3	3
Marcy mill:		
Installing feed end liners	8	5
Installing main liners	24 to 36	8
Installing grate bars	16	5
Hardinge mills:		
Renewing 20 pieces of liners and bars	8	5
Ball-tube mills:		
Renewal of feed end liners	6	4
Main liners	12 to 16	8
Screens	6	4
Minerals Separation machines:		
Changing impellers	16	4
Changing liners	16 to 20	6
Pumps:		
Changing liners	6	2
Changing liners and impellers	8	2
Filter:		
Changing cloth	12	6
Changing vacuum pipes	30	6

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COSTS

Table 9 presents a summary of concentrator costs for the year 1929, and Table 10 gives concentrator costs in units of power and labor for the same period.

Table 9.--Summary of concentrator costs for the year 1929

	Operating				Repairs		Total
	Labor	Power[1]	Supplies	Reagents	Labor	Supplies	
Breaking	$0.0380	$0.0179	$0.0065	-	$0.0093	$0.0139	$0.0856
Intermediate crushing	0.0043	0.0094	0.0004	-	0.0037	0.0097	0.0275
Primary grinding	0.0203	0.0544	0.0469	-	0.0058	0.0254	0.1528
Secondary grinding	0.0173	0.0351	0.0318	-	0.0033	0.0063	0.0938
Classifying, screening, and conveying	0.0076	0.0064	0.0002	-	0.0110	0.0134	0.0386
Flotation	0.0224	0.0289	0.0002	$0.0329	0.0036	0.0052	0.0932
Dewatering and handling concentrates	0.0164	0.0035	0.0116	-	0.0010	0.0004	0.0329
Assaying and sampling	0.0029	0.0001	0.0081	-	0.0003	0.0003	0.0117
Tailings disposal	0.0317	0.0014	0.0051	0.0149	0.0046	0.0006	0.0583
Water supply	0.0013	0.0001	-	-	-	-	0.0014
Miscellaneous	0.0093	0.0006	0.0464	-	-	-	0.0563
Totals	$0.1715	$0.1578	$0.1572	$0.0478	$0.0426	$0.0752	$0.6521

1 - Cost per kilowatt-hour including power generated locally = $0.007909.

Table 10.--Summary of concentrator costs in units of labor and power

	Labor			Power	
	Man shifts of 8 hours per ton of ore concentrated			Kw. h. per ton	Per cent of total power
	Operation	Repairs	Total		
Breaking	0.00822	0.00189	0.01011	2.258	11.32
Intermediate crushing	0.00085	0.00076	0.00161	1.194	5.98
Primary grinding	0.00405	0.00118	0.00523	6.893	34.55
Secondary grinding	0.00346	0.00069	0.00415	4.441	22.26
Classifying, screening, and conveying	0.00151	0.00225	0.00376	0.811	4.07
Flotation	0.00427	0.00075	0.00502	3.652	18.30
Dewatering and handling concentrates	0.00376	0.00020	0.00396	0.442	2.21
Assaying and sampling	0.00061	0.00005	0.00066	-	-
Tailings disposal	0.00682	0.00093	0.00775	0.171	0.86
Water supply	0.00029	-	0.00029	-	-
Miscellaneous	0.00260	-	0.00260	0.088	0.45
Totals	0.03644	0.00870	0.04514	19.950	100.00

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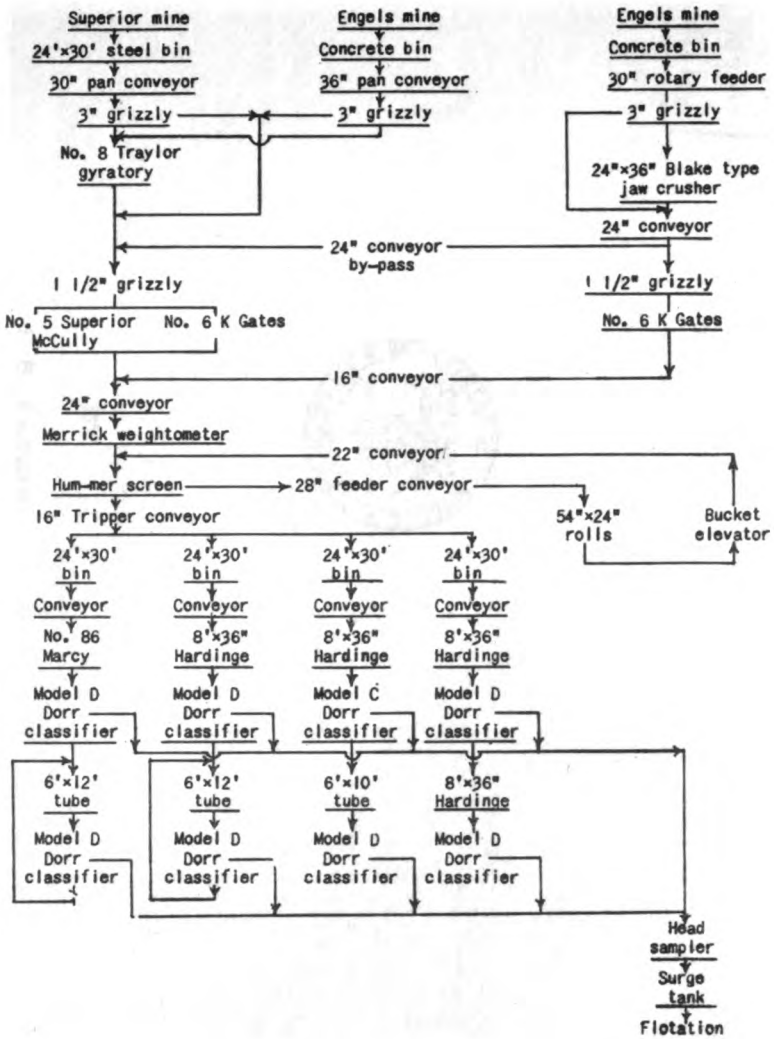

Figure 3.- Flow sheet of crushing and grinding operations

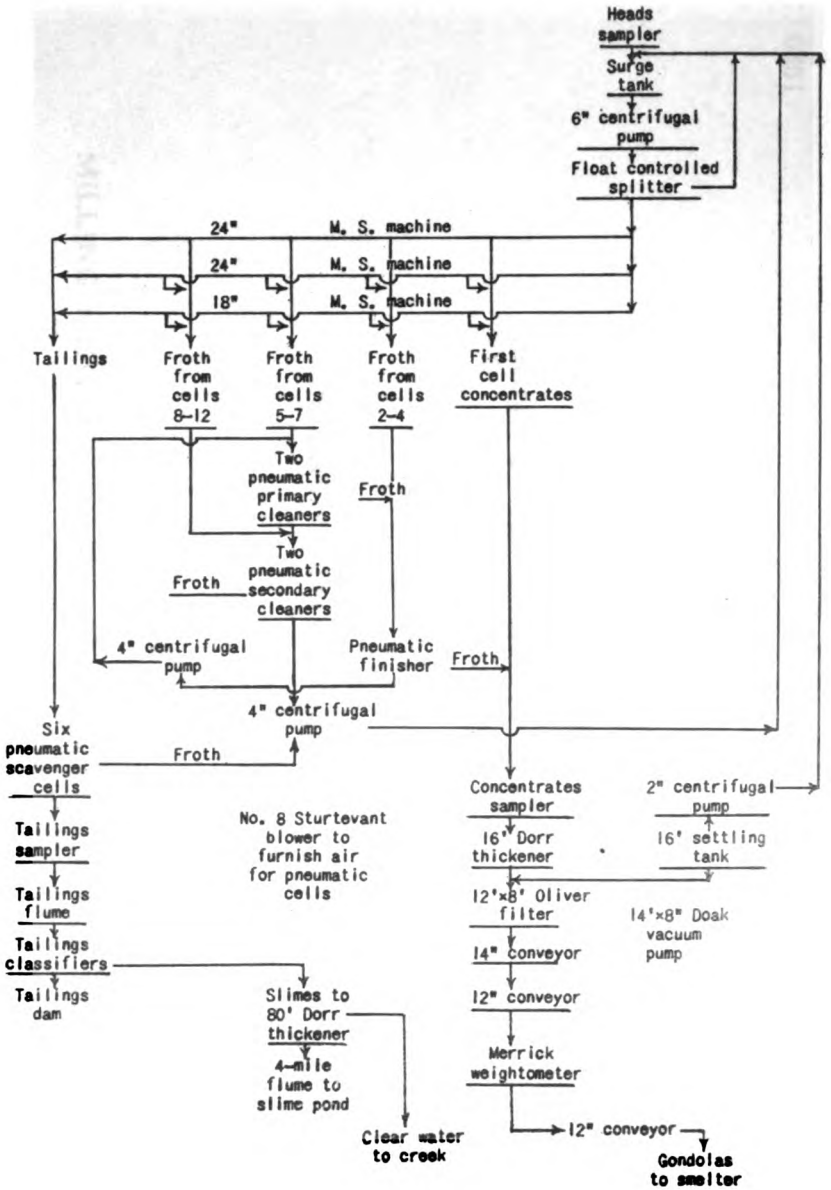

Figure 4.- Flow sheet of flotation and dewatering treatment

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